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COLUMN FLOTATION OF SYLVINITE

by

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COLUMN FLOTATION

OF

SYLVINITE

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SUMMARY

Column flotation is one of the most important innovations in flotation technology to occur in the last few years. Column flotation machines differ from conventional flotation machines in that the height to diameter ratio is much higher and they have no moving parts. Thus they are easier to maintain and are more energy efficient. Further, their mode of operation makes them better suited for producing high grade concentrates. Although work in this area began in the early 1960's it is only since 1982 that column flotation has started to gain acceptance. Canada is probably the major user of the technology and most of the applications have been in the sulphide industry. The single largest application is probably in the molybdenum cleaning circuits at Les Mines Gaspé where up to 13 stages of cleaning have been replaced by one or two stages of columns.

Coal is probably the second most important area in which column flotation is being tested. The application of column flotation to non-sulphide minerals has not been very extensive. The literature identified only one specific reference to the use of columns for potash flotation.

The theoretical aspects of column flotation e.g. scale-up factors and computer simulation equations are based on work done by Finch and Dobby since 1984. Factors such as reaction rate, rate constants, dispersion coefficients, vessel dispersion numbers and collection efficiencies have been identified and related to each other to provide a framework for relating laboratory test data to full scale equipment requirements and expected plant results.

For this project testwork was carried out at two potash operations in the Sussex area of New Brunswick. Both operations were investigating whether the use of column flotation would enhance the recovery of fine potash. The results from Plant A generally showed that using the column gave at least as good K_2O grades and approximately half the insol level. The results from Plant B indicate that column flotation gives better grades and recoveries than the conventional cells.

RESUME

La flottation sur colonnes est une innovation plus importante en la technologie de flottation. En la flottation sur colonnes la proportion de hauteur a diamètre de la machine est plus grand que flottation conventionnel. Les machines n'ont pas les pièces mobiles, ont plus faciles maintenir et utilisent l'energie plus bien. De plus, elles ont meilleures pour la production de concentrés de première qualité.

C'est seulement depuis 1982 que la flottation sur colonnes est accepté. Probablement, plus d'applications sont en Canada et en les opérations pour sulfides.

La deuxième domaine d'importance pour flottation sur colonnes est la flottation de charbon. Il y avais seulement un référence pour la flottation sur colonnes à utiliser en flottation de potasse.

La théorie de la flottation sur colonnes est basé sur la travail depuis 1984 de Finch et Dobby. Ils ont développé équations qué établir un rapport entre les variables.

Pour cet projet, étudier la flottation sur colonnes en la flottation de potasse fin, les essais faisont en deux usines en la région de Sussex. Les résultats indiquent que la flottation en colonnes est égal ou meilieur que la flotation conventional.

1.0 INTRODUCTION

Column flotation is a recent concept in mineral processing which made its start in copper-molybdenum operations and has since expanded almost exponentially to include all sulphide minerals and coal. Industrial minerals appear to have largely ignored column flotation. A literature search identified one paper which mentioned the use of columns for potash flotation. Verbal communication indicates that other work has been done but it seems that none of it has been published.

This project originated as a result of a request from Denison-Potacan Potash Corporation to use the laboratory flotation column owned by the Minerals Development Branch of the Department of Natural Resources and Energy. The project was followed up by RPC and a joint venture project between RPC, DNRE and DPPC was commenced under the Canada/New Brunswick Mineral Development Agreement. RPC and DNRE provided funding and RPC and DPPC provided personnel. As well, DPPC covered all of the required assaying. The project was expanded later to include PCA in a role similar to DPPC's.

The objectives of the project were to determine whether column flotation is a viable option to improve the recovery of fine potash and whether column flotation can increase the grade of the slimes concentrate. The scope of the project included a literature review; testwork to enable RPC personnel to become familiar with the cell operation; a review of previous work using the cell; and in-plant testwork at DPCC and PCA.

2.0 LITERATURE REVIEW

2.1 THEORY

The use of flotation columns is a relatively recent concept. Column flotation was invented by Pierre Boutin in the early 1960's (Wheeler, 1988) and was refined by Boutin, Tremblay and Wheeler. The first commercial use, however, did not occur until 1981 when Les Mines Gaspé, Quebec installed columns as the molybdenum cleaning circuit. Three columns replaced 13 conventional cleaner circuits and produced better results (Coffin and Miszczak, 1982).

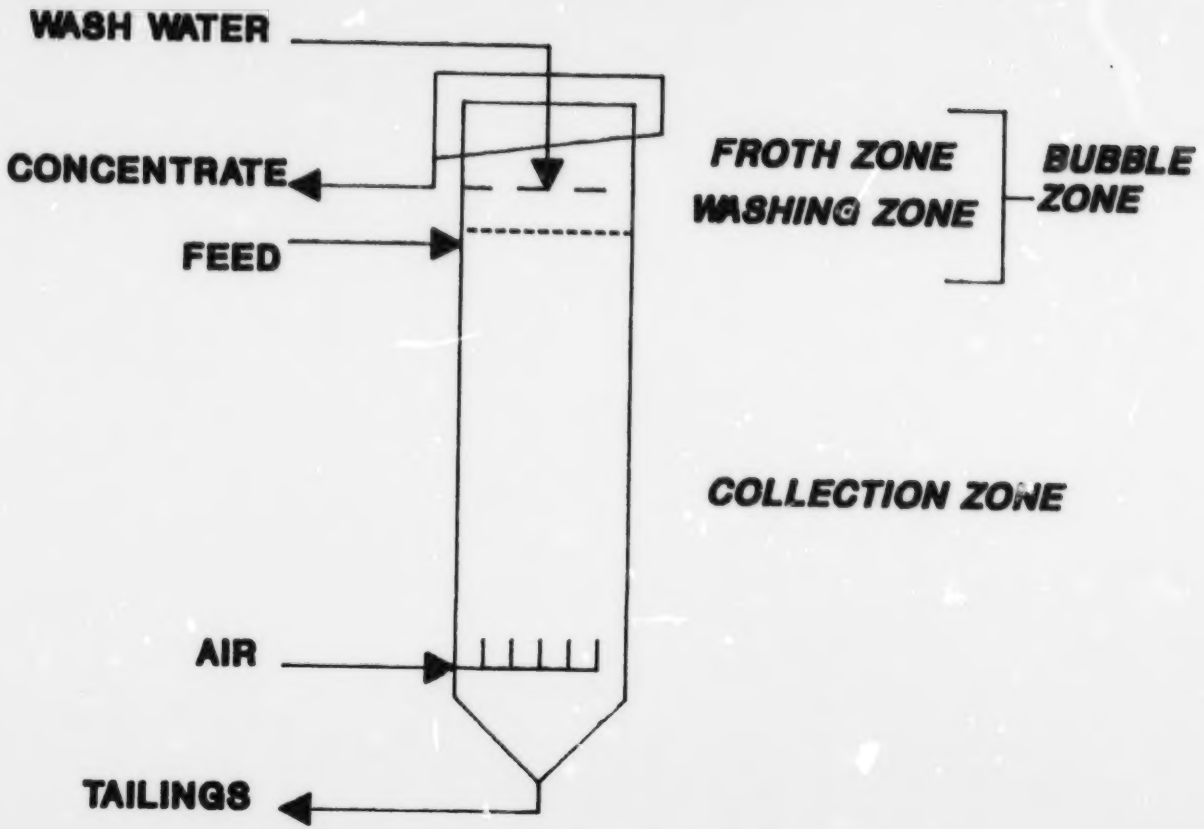
Boutin, Tremblay and Wheeler's early work was carried out in a 2 inch diameter laboratory unit and they rapidly progressed to a 12 inch diameter unit. Unfortunately, the Iron Ore Company of Canada (IOCC), where this work was being carried out, made a decision not to use flotation. Therefore all flotation testwork ceased at IOCC (Wheeler, 1988).

Although testing continued on sulphide ores, progress was slow. A number of factors influenced this including lack of capital which meant that sufficient testwork to interest the mining industry could not be carried out. During this time, however, a successful trial was carried out at Opemiska.

Following the successful application of flotation columns at Gaspé, their use has expanded considerably to include other base metals and coal operations.

Column flotation units differ considerably in design and operation from conventional flotation machines. The main design differences are that flotation columns have a high height to diameter ratio and they have no moving parts. A column flotation cell is illustrated in Figure 1. Column height can be up to approximately 13 metres and the column width can range from 0.3 to 2.5 metres. The cross-sectional area can be either circular or square. (Columns in China have been used which have a circular lower portion and a square upper portion (Weibai and Guoming, 1988).)

FIGURE 1
SCHEMATIC OF FLOTATION COLUMN



Two zones can be identified: a collection zone and a bubble zone. Depending on the wash water addition point, the bubble zone can be considered as two zones: a washing zone below the water addition point and a froth zone (the same as on a conventional flotation machine) above the wash water addition point. The bubble zone occurs above the feed entry point and the collection zone occurs below the feed entry point.

The feed entry point is usually one third to one quarter of the height of the column from the top. The conditioned feed slurry on entering the column moves down by gravity towards the tailing outlet. The feed particles contact rising air bubbles and those particles which adhere to the bubbles are transported into the bubble zone. In this zone, the bubble particle mixture is washed by a downward flow of fresh water referred to as the "bias". Weakly adhering and entrained gangue particles are washed away while the strongly attached particles flow upwards and are removed from the top of the column. The addition of wash water effectively means that very little or no process water reports to the concentrate. The concentrate overflows the top of the column into a launder and is removed for further processing.

2.1.1 MIXING

The two extremes of mixing in a reactor are plug flow and complete mixing. Plug flow is where the residence time of all the elements is the same while complete mixing exhibits a range of retention times from zero to infinity and the concentration is the same throughout the reactor. For a first-order rate reactor system exhibiting plug flow and a retention time t:

$$R = 1 - \exp(-kt) \qquad (1)$$

For a completely mixed system with a mean residence time of τ

$$R = \frac{K\tau}{1 + K\tau} \qquad (2)$$

where R = reaction rate

K = rate constant

Flow in a laboratory column approximates plug flow while plant scale columns exhibit a flow pattern between the two extremes. Addition of a tracer at the top of a recovery zone of length L gives the following mass transport equation to describe its concentration C at an axial distance x downstream from the addition point at time t:

$$D\frac{d^2}{dx^2} - U_i\frac{dc}{dx} - \frac{dc}{dt} = 0 \qquad (3)$$

D = axial dispersion coefficient (length^2/time)

U_i = interstitial velocity of the liquid or particle

A residence time distribution (RTD) of the solid or liquid is obtained by measuring the tracer concentration over time in the tailings. The RTD can be obtained theoretically if the mean residence time and the dimensionless vessel dispersion number N_d are known. N_d is given by

$$N_d = D/U_i\ L \qquad (4)$$

Equation 2 has been solved analytically (Wiehner and Wilhelm, 1956) for first-order reactions and any kind of entrance and exit conditions. The solution is:

$$R = 1 - \frac{4\ a\ \exp\ (1/2N_d)}{(1+a)^2\ \exp\ (a/2N_d) - (1-a)^2\ \exp\ (-a/2N_d)} \qquad (5)$$

Mixing parameters are measured so that their effect upon recovery can be quantified.

2.1.2 AIR ADDITION

In a mechanical flotation cell, the air is drawn into the cell by the rotation of the impeller. The mixing action of the impeller breaks down the incoming air stream into small air bubbles which are stable in the presence of a frother. In a column there is no impeller, therefore, the bubbles have to be introduced. This is achieved either with a sparger (usually a perforated tube) or by adding the air under pressure to an incoming stream of water.

A flotation column provides much more time for bubble-particle interactions. In mechanical flotation cells bubble-particle attachment occurs mainly in the turbulent hydrodynamic conditions of the zone around the impeller. These conditions can be detrimental in that these forces can pull weakly held particles off bubbles. In a flotation column the particles and air bubbles can interact in reasonably quiescent conditions over two thirds to three quarters of the length of the column.

An increase in the air flow rate does affect the flotation in that as the air rate increases the dispersion coefficient increases. Also, the collection rate constant is effected. When considering a gas bubble rising through a downward flowing slurry stream, the collection efficiency, E_K, of the bubble is defined as the fraction of those particles contacted by the bubble which adhere until the bubble reaches the cleaning zone. E_K is a complex function effected by many parameters including particle diameter, bubble diameter and particle hydrophobicity. A method relating the various parameters to collection efficiency has been developed by G. Dobby (1984).

E_K is directly related to K, the collection rate constant. The rate of collection of particles can be described by (Jameson et al, 1977):

$$\frac{dC_p}{dt} = (1.5\ V_g\ E_K/d_b)Cp \qquad (6)$$

Where C_p = particle concentration
V_g = superficial gas velocity
d_b = average bubble diameter

The rate constant $K = (1.5\ V_g E_K/d_b)$ (7)

Ultimately it is the fractional recovery of the valuable mineral, R, which is required. Recovery is determined by three factors: the rate constant, the mean residence time of the particles in the column, and the mixing conditions in the collection zone. Test work at Mount Isa Mines has shown that columns require much larger residence times than mechanical cells due to a lower rate constant. This may be because of the low kinetic energy of particles in columns.

2.1.3 CLEANING ZONE

The complex interactions of the cleaning zone are not well understood. Factors such as a gas rate, bias, solids content and wash water penetration into the cleaning zone all affect mineral recovery. Tests with tracers have shown that the bubble bed is very effect. e in removing gangue particles from bubbles. These tracer tests have shown also that less than 1% of the process water ends up with the concentrate. This implies that less than 1% of the hydrophillic particles report to the concentrate.

The downward flow of the wash water minimizes bubble coalescence thereby minimizing the amount of material that should be recovered but ends up back in the collection zone. Fortunately this material has, theoretically, as much chance of being recollected as incoming feed has of being collected.

2.1.4 SCALE-UP

Scale-up methods using a computer simulator have been devised by Del Villar and co-workers (1988). This is based on an earlier scale-up model of Dobby and Finch (1986). The simulator requires two experimentally determined parameters: carrying capacity and flotation rate constants.

The carrying capacity is determined by running the column with different feed percent solids while keeping the residence time constant. Timed samples of feed, concentrate and tailings are collected. A plot of concentrate solids flowrate in g/min versus feed solids flow rate in g/min is produced. A typical curve will show a maximum concentrate flow rate which when divided by the column cross-sectional area gives the carrying capacity in g/min/cm^2.

The flotation rate constants are determined by a second set of column flotation tests. A number of runs are made at different nominal liquid residence times which are the ratios of the volumes of the collection zone to the tailings flow rates. Recovery versus time graphs can be constructed for each mineral and the rate constants can be obtained from the slope of the plot.

Other parameters involved in the simulator include collection zone recoveries, particle mean residence time, vessel dispersion number N_p and final recovery. Collection zone recoveries can be calculated using equation 5, while particle mean residence time τ_p is given by

$$\tau_p - \tau_l \frac{U_l}{U_l + U_p} \qquad (8)$$

and N_p is given by equation 4 and

$$D_1 - D_p - 0.63d_c(V_g/1.6)^{0}.3 \ m^2/s \qquad (9)$$

Errors in the simulator predictions can be introduced by the following factors: variances from plug flow (assumed for rate constants), differences in feed characteristics between the "real" feed and the "tested" feed, particle settling velocity (for large and/or dense particles correction factors may be required), gas hold up in the collection zone which shortens the true liquid residence time, gangue rate constants which are not measured, carrying capacity in that the values are strongly dependent on the stream characteristics which can vary day to day, and rate constants in that although the equations are assumed to be first order, first order kinetics do not fit the data.

More information pertinent to scale-up is provided by R. Crozier (1990). He provides operating conditions in terms of superficial velocities (J) of the fluids. Typical values are as follows:

J_g (gas velocity)	=	0.5 - 3.0 cm/second
J_p (pulp feed velocity)	=	0.7 - 2.0 cm/second
J_w (wash water velocity)	=	0.1 - 0.8 cm/second
J_b (bias water velocity)	=	0.07 - 0.3 cm/second

When scaling-up the gas velocity is normalized for different column heights by the pressure correction:

$$J_g - \frac{P_c \, J_g^* \, \ln (P_T/P_c)}{P_T - P_c}$$

where J_g^* = standard gas velocity at the top of the column

P_c = absolute pressure at the top of the column

P_T = absolute pressure at the sparger

For a 10 metre column, P_T is approximately twice P_c therefore J_g approximately equals 0.69 J_g*.

2.2 PRACTICE

H.S. Soto (1989) reported some work he has carried out using "negative" bias in column flotation of coarse potash. His technique eliminates the use of wash water and, instead, introduces an upward flow of water at the bottom of the column. Apparently, this accelerates the recovery of the desired mineral and enables much coarser sizes than usual to be recovered. Using a feed containing approximately 70% plus 1.2mm material, he obtained the following results:

	COLUMN		CELL	
	Grade, K_2O	Recovery	Grade, K_2O	Recovery
Concentrate	60.2	65.3	59.4	32.8
Tail	12.1	34.7	21.2	67.2
Concentrate, +1.2mm	60.8	48.3		6.0

The recovery of + 1.2mm potash was significantly higher and the overall recovery was doubled when using the column versus traditional flotation.

Some of the more recent work by the U.S. Bureau of Mines - Salt Lake City Office was reported by J. McKay (1987). The USBM have investigated the processing of the Fish Creek fluorite deposit in Nevada. The feed contains 16% CaF_2, 66% SiO_2, 5% $CaCO_3$ and 0.2% BeO. The ore was crushed and screened at 297 μm (48 mesh) and the oversize reported to a rod mill. The -297 μm material (from the screen and rod mill) were conditioned at 20% solids with 1500 g/t Na_2CO_3, 1000 g/t Na_2SiO_3 and 500 g/t of 5% oleic acid in ethanol. pH was approximately 9.0. The rougher concentrate graded approximately 85% CaF_2 and was cleaned in a 12 foot column to produce a final concentrate grading 98% CaF_2.

The tailings were deslimed at 20 μm and the -20 μm material, which contained less than 4% of the CaF_2 and less than 1% of the BeO was discarded. The +20 μm material was repulped and conditioned with 2500 g/t H_2SO_4 (to give a pH <3) and 350 g/t of Arosurf MG98A (a primary amine) for mica flotation. The concentrate graded 90% mica at 90% recovery.

The tailings, which contained 95% of the BeO, were dewatered and conditioned with 2500 g/t HF (pH = 2.8) and 600 g/t of tallow amine acetate. A silicate concentrate which recovered 80% of the BeO at 1.8% and a pure SiO_2 tails were produced. The concentrate is passed through a magnetic separator to remove iron and treated with 600 g/t of hypochlorite to remove the tallow amine acetate. After washing, the acid is added and a beryl concentrate grading 5-7% BeO was produced.

In total USBM have investigated 20 different mostly non-sulphide mineral systems. With columns they have obtained the following results:

chromite: a 41% Cr_2O_3 concentrate in one stage and 45% Cr_2O_3 at 90% recovery in two stages. No desliming was required.

manganese: were able to achieve similar results to conventional flotation in one stage versus three stages.

fluorite: could achieve a 99.6% CaF_2 grade at 97% recovery

mica: obtained a concentrate grading 90% at 90% recovery versus 85% grade and 75% recovery with conventional flotation.

In addition by using flotation columns they improved the silver recovery at Skagit and doubled the PGM concentrate grade on Stillwater's ore.

INCO-Thompson have tested column flotation for Cu-Ni cleaning and pentlandite cleaning (1987). They used a 0.9 metre diameter by 11.9 metre column and the following conditions:

gas velocity	2.2 cm/sec @ 0.85 m^3/min (30 SCFM)
gas hold-up	33%
retention time	8-9 minutes
washwater velocity	0.14-0.3 cm/sec
bias (Q_T/Q_c)	1.1
frother addition	1-2 mL/min (into washwater)

The column was not successful for pentlandite cleaning but gave 86% copper recovery versus 44% in conventional cells in the Cu-Ni cleaning. They used eight natural rubber covered spargers with approximately 19.4 holes/cm^2.

Problems were related to low gas requirements at high slurry densities which lead to gas bubbles coalescing, the need to get a stable froth and operation of the tailings valve (a mechanical rubber pinch valve worked well).

Brenda Mines have obtained results similar to many copper-molybdenum operations which have replaced conventional cells with flotation columns in the molybdenum cleaning circuit. They have replaced 11 stages of cleaning with two 0.86m diameter by 11.3m columns. Their spargers are four pipes with 6m holes covered with filter cloth. The wash water is injected 5 cm below the froth surface.

Cominco-Sullivan have found that they can produce 50% of the final zinc concentrate in one pass through a 2.4m diameter by 11.6m column (1987). The feed inlet is 2.4m from the top. The feed rate is 160 TPH at 45% solids.

Cominco-Magmont are using a 0.76m diameter by 6.1m column on the zinc regrind product (1987). With a feed of 1.25 TPH at 15 to 17% solids they achieved a 3 to 5% increase in zinc grade however the recovery was 80% versus 92% with conventional cells. MgO was reduced from 0.6% to 0.18%. They use USBM spargers at an air rate of 1.08 m^3/min (38 SCFM). The froth depth was 0.8m.

Highland Valley Copper have tested columns on the molybdenum cleaning circuit; the bulk copper-molybdenum precleaning circuit; and the copper-molybdenum rougher, cleaner and rougher separation circuits. They were successful on the first two and unsuccessful on the last three.

3.0 TESTWORK

The testwork for this project was carried out at potash operations in the Sussex area of New Brunswick. The testwork done so far has been preliminary and has been designed to determine whether column flotation is an option for the recovery of fine potash. The testwork has not yet reached a stage where scale-up factors have been investigated.

Practical problems related to undertaking testwork in an operating plant have delayed the testwork quite extensively. Also, familiarization with the flotation column and its operation was necessary prior to carrying out any meaningful testing.

The test cell is a 5 cm diameter by approximately 5 metre high clear plastic column. The unit consists of five 1 metre sections each with a feed/sample port. The top section has a launder to collect the concentrate and the bottom section has a tailings discharge port. Air was supplied from the plant as was wash brine, when required.

3.1 TESTWORK PROCEDURES

At Plant A the flotation column was erected next to the slimes flotation circuit. Feed to the column was obtained by syphoning material through a one inch hose from the first flotation cell. This gave a more consistent feed flow and facilitated reagentizing. Since the feed rate to the flotation column was more than it could process, a by-pass valve was used to regulate the feed. The tailing flow was controlled by an adjustable clamp on the discharge line.

To operate the flotation column, the feed rate was set by the by-pass valve, the tailings discharge rate was controlled by the clamp and the two were balanced to give the desired froth level. Reagents were added to the first flotation cell as required. Air flow rate and wash brine addition were controlled separately.

After the relevant independent variables were set, the flotation column was operated for a few minutes and then samples of the feed, concentrate and tails were collected, filtered and assayed for percent K_2O and percent insol. Also, concentrate samples from the slimes circuit were collected.

Some problems were encountered with the wash brine feed system. When adding a small volume of brine through small nozzle openings, the nozzles continually became blocked. A system, whereby the brine could overflow a weir and run via gravity to the top of the froth zone was found to work well. Additionally, having a vacuum filter close to the flotation column was advantageous in that the samples could be filtered before there was any significant temperature drop in the sample.

At Plant B, the flotation column was erected in a convenient area close to the conventional flotation circuit. Feed to the column was pumped from the conventional flotation circuit feed box. Flow rate was adjusted by controlling the pump speed. Control of the froth depth and the tailings flow rate was achieved by the adjustable clamp on the tailings discharge line. After the parameters had been set, the column was operated for a short time and then samples of the concentrate, tails and feed were collected, filtered and assayed for K_2O.

4.0 TESTWORK RESULTS

Variables investigated during the test programme included feed rate, feed pipe inlet position, wash brine flow rate, air flow rate and the use of reagents.

At Plant A, the test cell was initially set-up to be fed tailings from a slimes circuit. The low percent solids, however, made operation of the cell very difficult. Therefore, the feed source was changed to feed to the slimes circuit. This material assayed 24.1% K_2O and 1.0% insol and was 100% -48%μm. Parameters investigated in these tests were as follows:

feed inlet position	1.5m or 2.5m from the top
froth level	12.7cm or 74cm from the top
air addition	high or low
wash brine addition	yes or no
reagent addition	yes or no

The results for the concentrates from the tests conducted at Plant A are summarized in Table 1. Looking at air flow and keeping the other parameters constant indicates that there is basically no difference between the two rates with respect to K_2O recovery and grade. Keeping everything except froth level constant the deeper froth level gives better K_2O recoveries and grades at feed position 1 while the thinner froth level seems better at feed position 2. Average K_2O recoveries for feed position 1, high froth level versus low froth level are 61.2% and 48.2% respectively while the respective grades are 67.9% and 47.2% K_2O. For feed position 2 the respective recoveries and grades are 48.9% and 58.7% and 50.5% and 51.3%. These figures indicate that recoveries are a little higher using feed position 1 and grades are a little higher when using feed position 2. The use of wash brine does not appear to have much effect on recovery but does seem to improve the grade somewhat.

Comparisons of averages of the data with respect to K_2O recovery provide the following observations:

TABLE 1

RESULTS FOR CONCENTRATES OF COLUMN FLOTATION TESTS AT PLANT A

Feed Pipe Position	1	1	1	1	1	1	1	1	1	1	1	1
Wash Brine	NO	NO	NO	NO	NO	NO	NO	NO	YES	YES	YES	YES
Froth Level	1	1	1	2	2	2	2	2	1	1	2	2
Air Flow	HIGH	HIGH	LOW	HIGH	HIGH	LOW	LOW	LOW	HIGH	LOW	HIGH	LOW
Wt Recovery, %%	21.60	47.90	32.90	9.60	32.00	23.00	31.20	21.00	31.50	33.10	32.30	32.70
K20 Recovery, %	45.40	78.30	59.50	18.20	65.80	40.90	56.80	41.70	64.20	58.60	59.80	54.20
%K20	48.00	39.40	49.60	45.20	45.70	48.60	51.60	47.90	53.00	49.70	48.60	43.10
% Insol	1.46	.66	.93	.70	.63	.56	.61	1.14	.69	.89	.75	.79
Feed Pipe Position	2	2	2	2	2	2	2	2	2	2	2	2
Wash Brine	NO	NO	NO	NO	NO	NO	NO	YES	YES	YES	YES	YES
Froth Level	1	1	1	1	2	2	2	1	1	1	2	2
Air Flow	HIGH	HIGH	LOW	LOW	HIGH	LOW	LOW	HIGH	HIGH	LOW	HIGH	LOW
Wt Recovery, %	25.20	31.20	24.50	16.80	34.00	35.40	29.80	10.00	18.00	28.80	25.20	27.30
K20 Recovery, %	51.80	67.60	58.50	27.70	55.50	63.10	59.50	24.00	56.50	56.50	58.20	57.20
% K20	48.70	48.00	50.20	41.10	49.90	50.60	52.60	55.40	58.80	51.50	49.60	53.60
% Insol	1.03	.50	1.39	1.15	2.05	1.81	1.21	.67	.76	1.21	.89	1.09

	K_2O Recovery, %
feed position 1	53.6
feed position 2	53.0
feed position 1 with wash brine	59.2
feed position 1 without wash brine	50.8
feed position 2 with wash brine	50.5
feed position 2 without wash brine	54.8
feed position 1 with low froth	48.2
feed position 1 with high froth	61.2
feed position 2 with low froth	58.7
feed position 2 with high froth	48.9
feed position 1 with low air	52.0
feed position 1 with high air	55.3
feed position 2 with low air	53.8
feed position 2 with high air	52.3

Overall, there does not seem to be much difference between feed positions 1 and 2, however, comparison of the various parameters between their two settings seems to be the opposite for position 1 compared with position 2.

In the final tests frother was added and K_2O recovery increased to an average of 82%. Also, the grade improved slightly from 47.5% to 48.3% K_2O.

A comparison of the results with the column versus the plant's conventional flotation

cells showed that for 22 of the 28 tests, concentrate grade from the column was higher than the concentrate grade off the conventional cells. Tails assays also tended to be higher for the column. The average of the tails for the 28 column tests was 15.4% K_2O versus the 9.9% K_2O average for 13 conventional tails assays.

The average insol grade for the conventional cells was 1.6% where as for the column the averages were 0.97% insol at feed position 1 and 1.15% insol at feed position 2. The column consistently produced a concentrate with a lower insol level.

At plant B, feed to the column was from the feed to the fines circuit. This material assayed 32% K_2O (average) and was 50% -53μm. The results of the preliminary tests are given in Table 2.

A comparison of Tests 3, 4 and 5, where the conditions were identical, shows a high correlation between recovery and feed grade. A comparison of Tests 6, 7 and 8, where the head grades are quite close and only the air pressure varied, shows recovery of 81.9% was highest with the highest air pressure but grade was highest at 55.2% with the lowest air pressure. The K_2O recovery in Test 2 versus Test 8 indicates that the shorter froth depth (0.7m versus 1m) produces better recoveries. Test 1, 3, 4 and 5 indicate that either a high amine dosage and/or the lower tailings flow rate are detrimental to the potash grade and/or recovery.

TABLE 2

RESULTS OF PRELIMINARY TESTS

	% K_2O				Operating Conditions			
Test	Feed	Concentrate	Tails	Recovery, %	A Mine g/t	Froth depth, m	Air Pressure Kg/m^2	Tailings Discharge L/min
1	32.0	53.2	24.2	44.7	245	1	7031	3
2	32.0	53.7	15.2	73.2	25	1	10547	4
3	25.9	46.4	15.5	60.3	150	1	10547	3
4	30.1	46.7	14.5	75.3	150	1	10547	3
5	25.7	47.3	15.3	63.1	150	1	10547	3
6	37.3	55.2	18.3	76.2	25	0.7	3516	4
7	35.2	53.1	16.9	76.3	25	0.7	7031	4
8	35.1	52.7	14.0	81.9	25	0.7	10547	4

5.0 CONCLUSIONS AND RECOMMENDATIONS

The results obtained in the limited testing conducted at both plants indicate that column flotation may give better grades and recoveries with lower insol levels when compared with conventional flotation. Such results would be beneficial in that with less insol and NaCL and more K_2O, the crystallizers would be able to operate more efficiently and thus more economically.

More testwork needs to be carried out to provide further optimization of the various parameters. Once these have been obtained, testing can be conducted to provide suitable data for calculating the various scale-up equations. In addition, testwork should be carried out to optimise the conventional circuits.

Once one circuit has been completely examined, other circuits in the plants can be evaluated following the same procedure as used for the fines circuits.

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